Pooley

ILLINOIS
COAL MINING INVESTIGATIONS
CO-OPERATIVE AGREEMENT

State Geological Survey
Department of Mining Engineering, University of Illinois
U. S. Bureau of Mines.

BULLETIN 2

Coal Mining Practice

IN

District VIII (Danville)

BY

S. O. ANDROS

Field Work by R. Y. Williams and S. O. Andros

ILLINOIS
COAL MINING INVESTIGATIONS
URBANA
1914

The Forty-seventh General Assembly of the State of Illinois, with a view of conserving the lives of the mine workers and the mineral resources of the State, authorized an investigation of the coal resources and mining practices of Illinois by the Department of Mining Engineering of the University of Illinois and the State Geological Survey in co-operation with the United States Bureau of Mines. A co-operative agreement was approved by the Secretary of the Interior and by representatives of the State of Illinois.

The direction of this investigation is vested in the Director of the United States Bureau of Mines, the Director of the State Geological Survey, and the Head of the Department of Mining Engineering, University of Illinois, who jointly determine the methods to be employed in the conduct of the work and exercise general editorial supervision over the publication of the results, but each party to the agreement directs the work of its agents in carrying on the investigation thus mutually agreed on.

The reports of the investigation are issued in the form of bulletins, either by the State Geological Survey, the Department of Mining Engineering, University of Illinois, or the United States Bureau of Mines. For copies of the bulletins issued by the State and for information about the work, address Coal Mining Investigations, University of Illinois, Urbana, Ill. For bulletins issued by the United States Bureau of Mines, address Director, United States Bureau of Mines, Washington, D.C.

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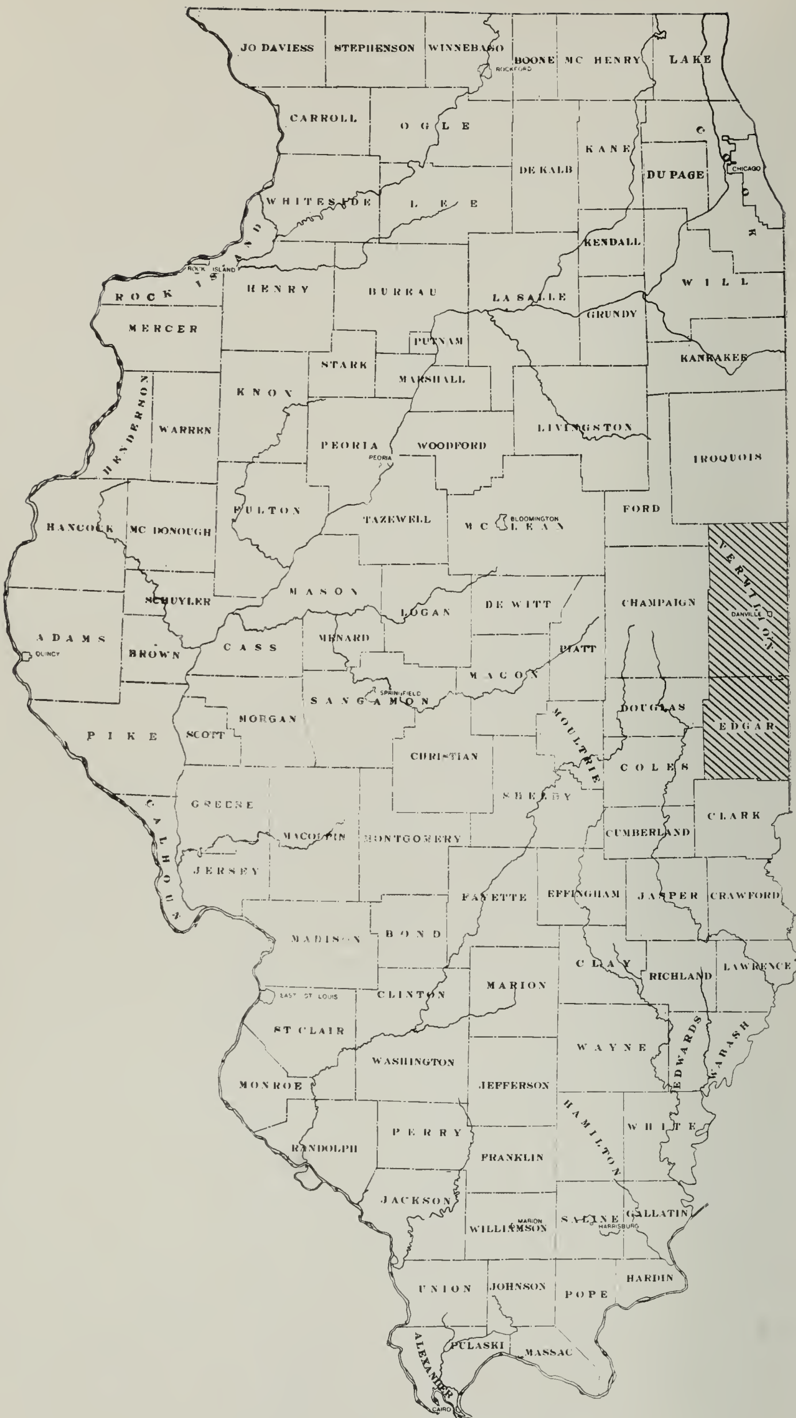

FIG. 1. Map showing by cross-hatching the area of District VIII

COAL MINING PRACTICE IN DISTRICT VIII (DANVILLE)

By S. O. ANDROS
Field Work by R. Y. Williams and S. O. Andros

INTRODUCTION

The Danville district, including Vermilion and Edgar counties, as shown in Fig. 1, is District VIII of the Illinois Coal Mining Investigations. It contains mines working in beds 6 and 7 of the Illinois State Geological Survey correlation.

A detailed description of the districts into which the State has been divided and the method of collecting the data upon which this bulletin is based is contained in Bulletin 1, "A Preliminary Report on Organization and Method."

Vermilion County at present ranks eighth among the fifty-two coal producing counties of the State, its total output for the year ending June 30, 1912, being 3,374,443 tons, which amount is 5.8 per cent of the total output of the State. Of this amount 20.2 per cent, 683,789 tons, were mined by machines. The total tonnage was produced by forty-nine mines with 4,007 employees, or 5.0 per cent of the total employees of the coal mining industry in Illinois. Of these forty-nine mines, eighteen were shipping and thirty-one local.

Edgar County produced a small amount of coal in 1911, but ceased production during that year and in 1912 was not a contributor to the production of the district.

TABLE 1—*Statistics for District VIII for the Year Ending June 30, 1912**

	District VIII (all mines)	State (all mines)	Per cent of district
Total production	3,374,443	57,514,240	5.8
Average daily tonnage	18,339	359,464	....
Number tons mined by machines	683,789	25,550 019	2.7
Average days of active operation	184	160	....
Number days of work performed in 1912	737,288	12,705,760	5.8
Total employees	4,007	79,411	5.0
Number surface employees	305	7,049	4.3
Number underground employees	3,702	72,362	5.1
Number underground employees per each surface employee	12.2	11.3	....
Number tons mined per day per employee	4.6	4.5	....
Number tons mined per day per surface employee	60.1	50.9	....
Number tons mined per day per underground employee	4.9	4.9	....
Number fatal accidents	18	180	10.0
Per cent from falling coal or rock	77.7	54.4	....
Per cent from pit cars	5.5	18.8	....
Per cent from explosives	5.5	7.2	....
Number deaths per 1,000 employees	4.5	2.3	....
Number tons mined to each life lost	187,469	319,524	....
Number non-fatal accidents	59	800	7.4
Percent from falling coal or rock	47.5	45.5	....
Per cent from pit cars	27.1	26.3	....
Per cent from explosives	5.1	2.6	....
Number injuries per 1,000 employees	14.7	10.1	....
Number tons mined to each man injured	57,194	71,893	....

* Compiled from Thirty-first Annual Coal Report of Illinois.

Table 1 gives comparative statistics for the district and for the State for the year ending June 30, 1912.

Acknowledgments are due to the operators of the district who offered every facility for study of the mines and to the superintendents and mine managers who accompanied the engineers through the underground workings. Valuable assistance was rendered by Mr. Thomas Moses, and Mr. W. S. Burris, superintendents of the Bunsen Coal Company in reviewing the manuscript of this bulletin.

HISTORY

Mining on a commercial scale began in 1866 when Wm. Kirkland and Hugh Blankeney and Mr. Graves opened a stripping mine on Grape Creek. In 1870 considerable work was done at the West Vermilion Heights shaft, and through the early seventies the Grape Creek Coal Company opened up what are now known as the Old Grape Creek mines, about four miles southeast of Danville, midway between Danville and Westville on the line of the C. & E. I. R. R. This company went into the hands of a receiver in 1893 and its mines were foreclosed to the Eastern Illinois Coal Company, which leased its holdings to the Brookside Coal Company. Until 1892 it was not supposed that the Grape Creek or No. 6 bed extended under the prairie west of Danville and the first shaft in that territory was sunk by Michael Kelley in that year. This mine was known as Kelley No. 2, Mr. Kelley having previously sunk a shaft, Kelley No. 1, about five miles south and one mile east of Danville. In 1895 the Westville Coal Company sunk the Westville No. 1 shaft, which at that time was the opening farthest west and south in the No. 6 bed. This mine was rapidly developed by electric mining-machines and in 1898 was producing 2,000 tons of coal in eight hours of hoisting. This is supposed to be the first mine in Illinois to produce so large an amount of coal in an eight-hour shift. The Catlin mine, sunk in 1895 by A. C. Daniels, was sold to Jones and Adams, who, in turn, sold it to the Chicago Collieries Company. It is now being operated by the Danville Colleries Company in the Danville or No. 7 bed. This is the only mine in the district where beds 6 and 7 were both operated from the same shaft.

After 1895, mines of large production followed very rapidly. The first self-dumping cage in the district was installed at the Pawnee mine in 1898. The Kelley Coal Company in 1896 sank the Kelley No. 3 shaft, located one-half mile west of Westville, and in 1901 this mine produced the largest amount of coal of any mine in the Danville district. In the same year the Brookside Coal Company operated the Brookside No. 1, a shaft mine on the C. & E. I. R. R., one and one-half miles from Westville Station. This company also opened a small drift mine, Brookside No. 2, located one and one-half miles east of No. 1.

In 1899 the Himrod Coal Company sank the Himrod shaft and established the town of Himrod. The mine was equipped with mining-machines operated by compressed air, but was abandoned in 1907. In 1902 the Kelley Coal Company opened its No. 4 mine. In 1903 the Westville Coal Company sank its No. 2 shaft and in the same year the Electric Coal Company, W. G. Hartshorn, President, opened the Electric mine four and one-half miles west of Danville. In 1904 the Westville

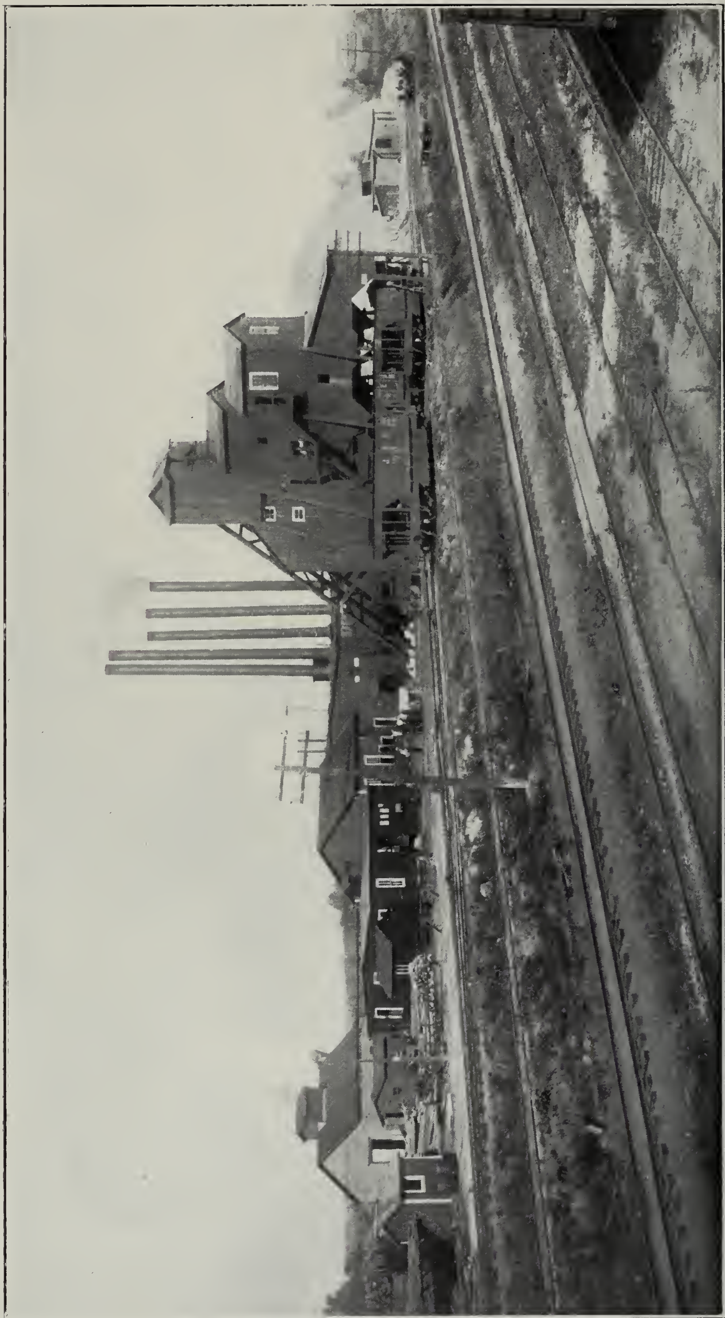

Fig. 2. Typical surface plant of a shaft mine

Coal Company sank the No. 3 and No. 4 shafts at Westville, the No. 4 mine being now (1913) the most westerly mine in operation in the district. The mines of the Westville Coal Company were taken over in 1905 by the Dering Coal Company.

The Little Vermilion shaft was sunk in 1905, about one and one-half miles north of Georgetown. In that year the properties of the Himrod Coal Company and the Kelley Coal Company were consolidated by the Illinois Traction System and in 1907 were sold to Mr. R. R. Hammond, who sold them in 1909 to the Bunsen Coal Company.

The Fairmount mine was opened by the Fairmount Coal Company in 1907 at Bennett Station, ten miles southeast of Danville.

A stripping mine at Missionfield, six miles west of Danville, was operated under a lease given by the Consolidated Coal Company, till its abandonment in 1900. Operations were again begun on this tract in December, 1909, by the Missionfield Coal Company. This company at first used the drag-line shovel in removing the overburden, but in 1910 purchased a revolving shovel which, at the time of the purchase, was the largest revolving steam-shovel in the world. The old salt-wells of an early period were located about 100 feet from the present tipple of this company, and it was from these salt-wells that the Salt Fork River derived its name.

Stripping operations were begun by the Western Brick Company in 1902 on land located two miles west of Danville where formerly there had been a room-and-pillar mine, and this company still operates a portion of its property on the room-and-pillar system. From the shale overlying the coal, this company has an annual output of eighty-five million brick.

DESCRIPTION OF COAL BEDS

In this district all the large outputs are produced from the No. 6 bed of the Illinois State Geological Survey correlation.

The chief characteristic of the No. 6 bed which averages 6 feet in thickness, is the presence of a blue-band which divides it into upper and lower benches. This blue-band varies from soft dust to hard gray shale and occurs about 2 feet above the floor. In addition to this blue-band there are several shale and sulphur-bands of variable horizontal and vertical extent.

The roof over coal No. 6 is variable. Near Danville the immediate roof is a grayish black shale about 6 feet thick. This shale, lying between the coal and the cap-rock of dark gray nodular limestone makes an easily supported roof. In the vicinity of Westville and Georgetown, the immediate roof is usually a gray shale which shows no distinct bedding, has little cohesion, falls in conchoidal masses, and is extremely difficult to support. Further, stringers of coal extend from the bed proper into the roof material and render the roof more difficult of support. In isolated cases there are 3 to 4 inches of black shale between the coal and the gray shale which forms the cap-rock. Whenever this black shale is broken, air and moisture disintegrate the gray shale cap-rock and the roof becomes insupportable.

In all parts of the Danville district the floor is a soft fireclay.

The No. 7 bed varies in thickness from 2½ to 5½ feet and averages 5 feet. The coal has two benches separated by a clay-band 1 inch thick, which persists through the bed from 6 to 8 inches above the floor. The two benches present no great difference in appearance or in physical character except locally where the top bench is harder and has a brighter luster. The No. 7 bed generally has slightly more impurities than the No. 6 bed, higher volatile matter, lower fixed carbon, and higher sulphur content as shown by Table 2, obtained from analyses of thirty-one face samples in No. 6 and of eighteen face samples in No. 7. A detailed discussion of these analyses will be given in a later bulletin. The bands of pyrites occur persistently at a height of 20 to 26 inches above the floor and "sulphur balls" or nodular concretions of pyrites are present in such quantity as to make profitable their separation from the coal by hand picking in the mine and by a further separation on the surface in rotating cylinders.

In both beds in the district there are numerous rolls, called "faults," or "horsebacks" by the miners. These rolls appear to have been due to unequal settling of the coaly matter and the necessary readjustment of the roof materials, during the formation of the coal. In many cases the roll entirely displaces the coal.

The mines in District VIII are shallow and the deepest mine does not exceed 300 feet. A detailed and comprehensive report on the geology of this district is in preparation and will be published later as a separate bulletin.

Table 2—*Comparative Analyses of Beds Six and Seven Made by Prof. S. W. Parr**

Bed	Coal moisture free				Moisture	B. t. u.	
	Volatile matter	Fixed carbon	Ash	Sulphur		Moisture free	Unit coal
Six	41.94	47.14	10.92	2.98	14.45	12,764	14,575
Seven	44.01	44.53	11.47	3.37	12.99	12,807	14,740

* Number of analyses averaged, coal No. 6, 31; coal No. 7, 18.

MINING PRACTICE

In the Danville district the beds lie at varying depths below the surface and various means of reaching the coal are employed. At eighteen of the forty-nine mines in the district the coal is reached by a

FIG 3. Slope lined with concrete

shaft; at seventeen by a drift; at ten by a slope; and at four the coal is exposed by removing the overburden with a steam-shovel. At one property, where the surface exhibits considerable relief, No. 7 is being worked through a slope and by a drift and a large area is also being stripped.

ROOM-AND-PILLAR SYSTEM

METHOD OF WORKING

Mining in the district was begun before the refinements of method necessitated by the present keen competition in the industry were known. Much progress has been made in efficiency and safety of mining in several mines but there still exists many examples of "gopher-mining" which are typical of the inefficient and unsafe methods of an earlier period. Six of the eighteen shipping mines of the district were examined and the ages of these mines varied from five to nine years.

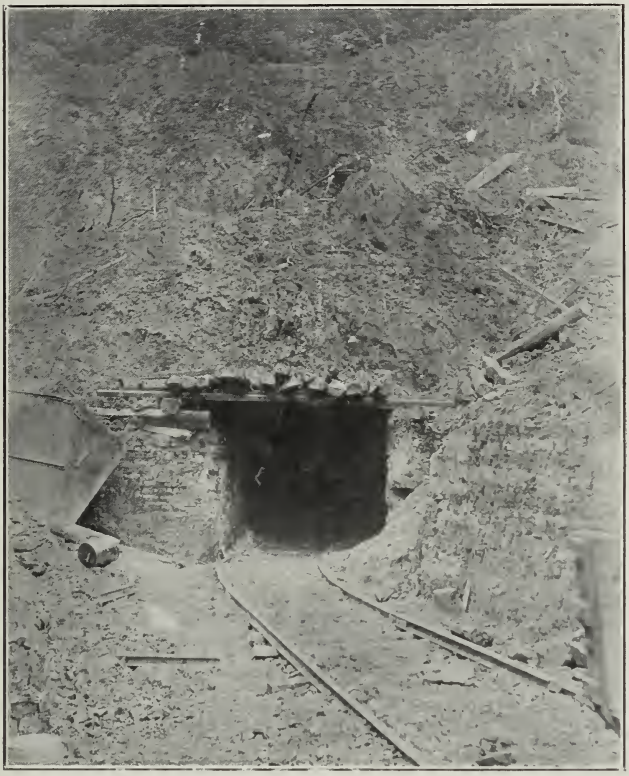

FIG 4. Portal of a drift mine

Those mines in the district which are now producing the greatest daily tonnage were not opened by the present operators but were projected by former owners when the comparatively small outputs did not demand rapid and uninterrupted haulage and hoisting. Even had the original projections been made according to a modern system, it would have been impossible to adhere to them on account of the many rolls in the roof which cause a deviation from an outlined system. Whether the

bed is reached by a shaft, Fig. 2, a slope, Fig. 3, or a drift, Fig. 4, the system of mining generally adopted in the district is the simplest form of double-entry room-and-pillar working. In each mine examined a main entry is driven towards the property lines from each side of the hoisting-shaft, paralleled by an air-course driven from the air-shaft. Along the main entry pairs of cross-entries are driven at a right angle from both sides of the main entry at intervals of 300 to 500 feet. Rooms

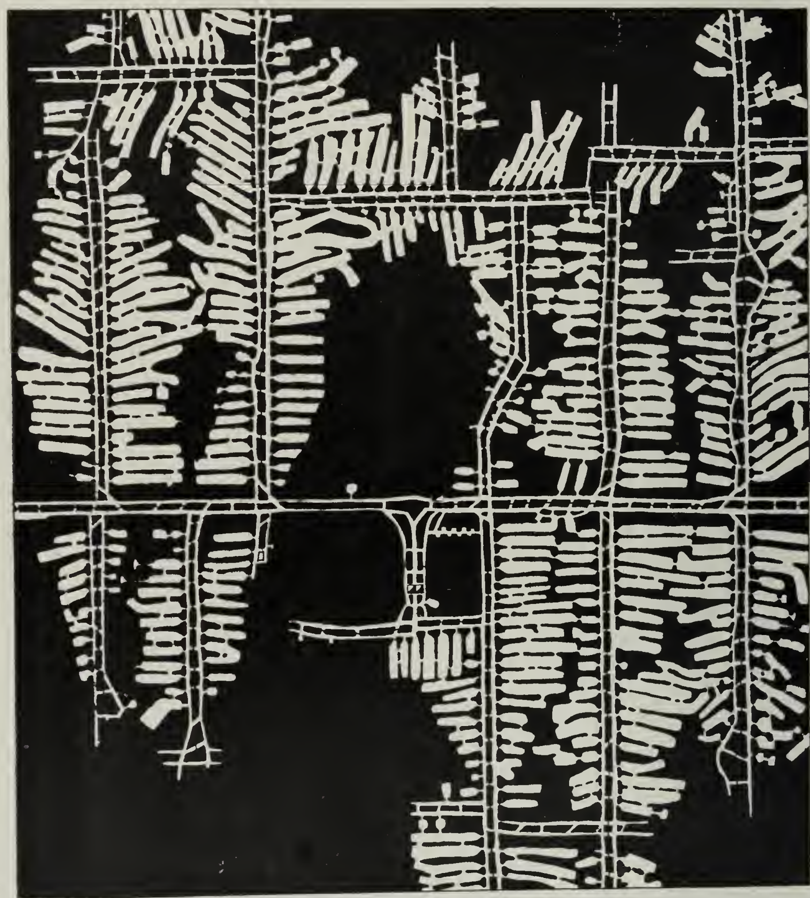

Fig 5. Plan of typical mine

are turned at a right angle from each entry of a pair of cross-entries. The roof is usually so difficult to support that both main and cross-entries are very narrow and room-necks are driven narrow and long to provide a large room-stub.

The frequent occurrence of rolls has a marked effect on the manner of driving rooms. In a roll area it is difficult to support the roof and

the expense of driving through the hard rock of the roll is great. Consequently, as shown in Fig. 5, which is the map of a mine typical of the district, when a roll is encountered in driving rooms it is customary to change the direction of the room and to drive it parallel to the roll until the coal resumes its normal condition. Often it is necessary to abandon a room before it has been driven its proper length. As the rolls are of frequent occurrence, the amount of coal that will be gained in any section of the mine is problematical. Consequently, the operator, on reaching that portion of the coal where the seam regains its normal thickness will attempt to get as much of the coal in the seam as possible during the first working. Little attempt is made to preserve a constant room-pillar width and the practice of gouging pillars is common in the smaller mines. No systematic pillar drawing is attempted because with present practice there is but little pillar left to draw when the rooms

FIG. 6. Surface subsidence near Westville

are driven up to their full length. The roof is so treacherous, especially in the vicinity of the rolls that it is not safe to leave wide spans of roof unsupported by pillars. The width of room-pillars at the mines examined varied from 4 to 16 feet and room-widths from 21 to 43 feet. Table 3 gives dimensions of workings at each mine examined. It will be noticed in this table that narrow room-pillars were found in Mine No. 91, where the following dimensions were measured:

Room centers, 47 feet.
Room widths, 43 feet.
Room-pillar width, 4 feet.

Although pillar-gouging in the district has resulted in a high percentage of extraction from the bed in the first working, it has caused a subsequent loss of coal through squeezes due to narrow pillars. The average extraction for the six mines examined is 70 per cent. Table 4

gives the per cent of bed extracted at each mine. These percentages were calculated from measurements made in the mine and were checked by figures of production per acre obtained from the books of each operating company and by planimeter measurements of mine maps.

TABLE 3—*Physical Characteristics of Mines*

Number mine	Name of bed	Depth of shaft	Pillar widths in feet				Entry widths in feet		Rooms in feet		Room—necks in feet		Widths of cross-cuts in feet.
			Main entry	Cross entry	Main barrier	Room	Main	Cross	Length	Width	Width	Length.	
Ninety-one......	6	217	25	21	40	4	9	9	200	43	9	19	5
Ninety-two......	6	240	35	30	60	3	7	7	240	24	9	18	9
Ninety-three	6	186	21	21	50	16	8	9	200	24	9	9	9
Ninety-four	7	90	16	12	17	6	7	6	240	24	9	9	9
Ninety-five......	6	90	......	......	......	11	6	6	500	25	9	12	9
Ninety-seven....	7	223	21	21	50	9	7	8	150	21	7	12	7

TABLE 4—*Percentage of Coal Extracted from Bed*

Mine number....................................	91	92	93	94	95	97
Per cent of extraction............................	55	68	75	81	82	68

Such a high percentage of extraction leaves large areas of roof unsupported by pillars and causes frequent squeezes. Few mines in the

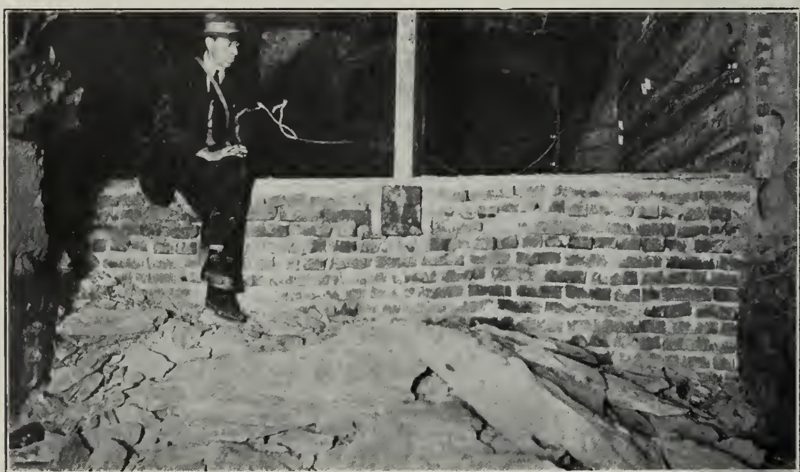

FIG. 7. Tongue-and-goove lining of overcast

district have been free from squeezes and in some instances the operating companies have become involved in litigation over damage to surface improvements through surface subsidence. Fig. 6 shows a surface sub-

sidence caused in 1904 by a squeeze in the mine underlying at a depth of 210 feet. The loss of coal by squeezes and the cost of re-driving entries through squeeze areas add to the cost of mining coal, yet in several mines narrow pillars and wide rooms were found, although the same dimensions had caused squeezes in the past.

More serious than the additional cost of mining is the loss of life due to unsafe working dimensions. The record of this district with respect to fatal and non-fatal injuries is bad. The miners of the district are largely of foreign birth; Italians, Lithuanians, and Poles pre-

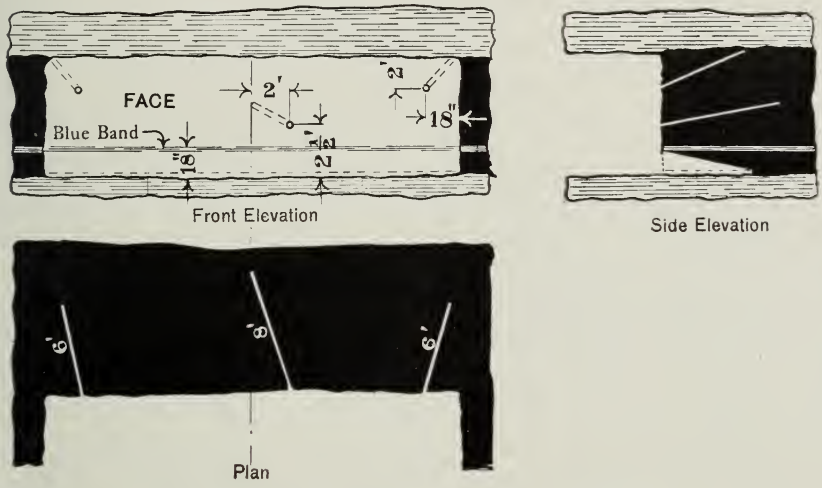

Fig. 8. Customary method of placing shots in undercut face

dominating, and it requires constant vigilance on the part of the operators to secure observance of the simplest rules for safe mining. The causes contributing to the excessive accident rate of the district are:

Treacherous roof.

Unsafe dimensions of working.

Lax discipline at the face.

More face-bosses with defined power to enforce safe propping of the roof at the working face would undoubtedly decrease the number of accidents.

Table 5 gives the comparative records for accidents of the State and the district for the years 1909, 1910, 1911, and 1912. It will be seen that in spite of the efforts of the operators to decrease the risk of mining by widening entries and by calling the attention of the men to the dangers of their occupation, no marked improvement has been brought about in the district as a whole. The miner will not take proper measures for his own protection unless he is compelled to do so and until defined power is given to face bosses the yearly death roll will retain approximately its present size. Table 6 gives an analysis of the causes of fatal and non-fatal accidents in District VIII and in all the other districts of the State combined for the year ending June 30, 1912:

TABLE 5—*Relation Between Accidents, Number of Employees, and Tonnage*

	State				District VIII			
	1909	1910	1911	1912	1909	1910	1911	1912
Number men killed	213	406	157	180	12	11	21	18
Number men seriously injured	894	742	709	800	56	35	39	59
Number tons mined to each man killed	230,816	120,000	319,523	319,524	185,136	184,860	142,207	187,469
Number tons mined to each man seriously injured	54,993	65,657	70,754	71,893	39,672	58,099	81,769	57,194
Number killed per 1,000 employees	2.9	5.4	2.0	2.3	3.7	3.3	5.7	4.5
Number injured per 1,00 employees	10.7	9.9	9.2	10.1	17.5	9.6	10.0	14.7

It will be seen from this table that although the per cent of non-fatal accidents from fall of rock or coal is only slightly in excess of the rate for all the other districts combined, the per cent of fatal injuries from this cause is excessively high. The desire of the present operating companies to prevent accidents is shown in several mines in the district where entries have been widened and retimbered to provide safe passageways, dangerous curves have been eased, and the roadbed thoroughly cleaned.

TABLE 6—*Causes of Accidents in District VIII and in All Other Districts Combined, 1912**

Cause	All other districts combined	District VIII
FATAL ACCIDENTS		
Per cent from falling coal or rock	51.8	77.7
Per cent from pit-cars	20.4	5.5
Per cent from explosives	7.4	5.5
NON-FATAL ACCIDENTS		
Per cent from falling coal or rock	45.3	47.5
Per cent from pit-cars	26.2	27.1
Per cent from explosives	2.6	5.1

* Thirty-first Annual Coal Report of Illinois.

The operators of the larger mines have taken precautionary measures against fire, and the fire protection at mine No. 93 is especially good. At this mine the method of conveying lubricating oil to the shaft-bottom is unusual in the district, and, indeed, in the State. On the surface at a distance of 100 feet from the hoisting shaft three oil tanks are sunk 5 feet deep in the ground. One tank of 400 gallons capacity contains black oil; one of 250 gallons capacity contains engine oil; and a third of 200 gallons capacity holds cylinder oil. Pipes from these tanks are carried down the pipe-way in the hoisting-shaft and the various oils are pumped direct to the bottom as needed. This method obviates the necessity of taking oil into the shaft in barrels or in cans and does away with storing oil in the run-around.

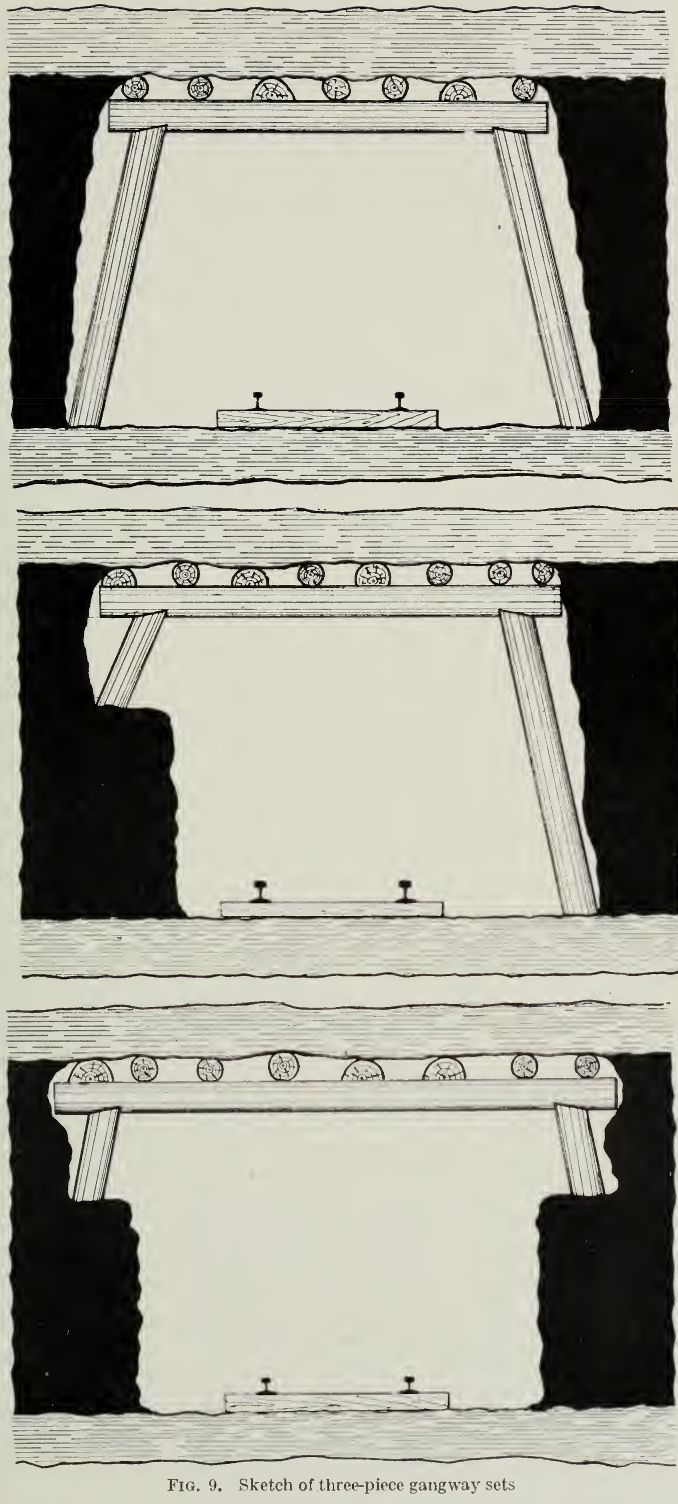

FIG. 9. Sketch of three-piece gangway sets

In the majority of mines in the district the mules are stabled underground. In some cases the hoisting-shaft is too small to permit daily hoisting of mules, but the fire-risk from underground stabling in the district is minimized as much as possible in the larger mines. The transportation of hay from the surface to the stables and its exposure in the mangers at feeding time is always a source of danger. Although the provisions of the State law in regard to transportation of hay and its storage underground, if followed, give reasonable security against fire from this source, many operators in the State prefer to stable their mules on the surface. In mine No. 52, located in Southern Illinois, it costs $0.000037 per ton of coal hoisted to lower each mule daily. The company has twenty-four mules underground and the total time consumed in lowering them is thirty minutes. The same time is required to hoist them. The total cost of lowering and hoisting twenty-four mules daily is $0.000888 per ton of coal hoisted. Fire-risk is thus avoided and the animals are kept in better condition.

Table 7—*Per Capita Production of Employees*

	Mine number						District in fiscal year 1912	All other districts combined in fiscal year 1912*
	91	92	93	94	95	97		
Average daily tonnage	1,250	150	2,600	800	50	300	18,339	311,021
Total number employees	265	40	620	230	9	78	4,007	72,634
Number surface employees	25	5	20	20	2	8	305	6,226
Number underground employees	240	35	600	210	7	70	3,702	66,408
Number miners, loaders and machine men	159	16	411	180	7	48	*2,506	50,812
Number underground employees per each surface employee	9.6	7.0	30.0	10.5	3.5	8.7	12.2	10.7
Tons of coal per day per each employee	4.7	3.8	4.2	3.5	5.6	3.8	4.6	4.3
Tons of coal per day per each surface employee	50	30	130	40	25	37.5	60.1	49.9
Tons of coal per day per each underground employee	5.2	4.3	4.3	3.9	7.1	4.3	4.9	4.7
Tons of coal per day per face worker (miners, loaders and machine men)	7.8	9.4	6.3	4.4	7.1	5.4	*6.0	6.1

* Shipping mines only.

The production of employees at each mine examined is shown in Table 7. The number of underground employees per each surface employee in the district is high compared with the number for the shipping mines of the other districts combined because several mines in District VIII have surface and general roustabout work well systematized.

VENTILATION

In the mines of large tonnage in this district the ventilation of the workings is excellent and the air supply is in excess of legal requirements. The air at the working face is as pure as mine air reasonably can be expected to be. Table 8 gives data on ventilating equipment for each mine examined.

Table 8—*Data on Ventilating Equipment*

No. of mine	Air-shaft				Fan			Material of fan-house
	Depth in feet	Size in clear in feet	Number compartments	Lining	Type	Diameter in feet	Width of blade in feet	
91	217	8 by 12	2	Timber	Capell	16	4	Timber
92	240	6 by 14	2	..do	..do	10	4	..do
93	186	8 by 12	2	One timber, one concrete	..do	24	7	..do
94	90	7 by 14	2	Timber	..do	12	6	..do
95	40	4½ by 4½	1	..do	Paddle	3	1½	..do
97	223	6 by 10	2	..do	Capell	14	6	..do

Explosive gas is seldom found in active workings, and even in abandoned workings is found only in small quantity. In the history of the district there have occurred a few minor gas explosions.

The coal of this district when air-dried and tested in the laboratory at Urbana is the most explosible of all the coals in the State, and in the rooms, the rib-dust when dry, and the dust in suspension in the air, are more explosible than the dust in rooms in other districts. The average pressure developed by the coal dust when tested in the explosibility apparatus at the laboratory at Urbana is compared with the pressures developed by the coal-dust of other districts in Table 9.

Table 9—*Pressures Developed by Face Samples in Explosibility Apparatus*

District	Number samples	Pressure in pounds per square inch at 2,192 °F
I	11	8.400
II	5	5.880
III	5	7.805
IV	17	7.700
V	7	7.105
VI	16	5.950
VII	24	7.175
VIII Danville	6	8.925
IX	3	8.505

Although the rib-dust of the rooms is very explosive when dry, it is not very explosible as found on the ribs because it has such a high moisture content. The moisture contained in rib-dust samples of District VIII as received at the laboratory in Urbana averages 25.31 per cent as compared with 12.24 per cent for all the other districts of the State. The rib-dust and road-dust of the haulage-ways contain a large amount of inert shale supplied through the grinding of the numerous roof-falls by car wheels and by the feet of men and mules. The ash in haulage-way rib-dusts averages 38.59 per cent for the Danville district. All rib-dusts in the district are constantly damp even in winter and the uniform seepage of surface water, due to the shallowness of the workings renews the mechanically-held moisture as fast as it is

evaporated by the ventilating current. Consequently the relative humidity of the mine air is high, both in the return air current and in the rooms. Table 10 gives readings taken with a sling psychrometer on the surface and in the return air-current and at the working face.

TABLE 10—*Humidity Data for District VIII*

No. of mine	Date	Surface readings		Readings in rooms		Readings in return air-course		Remarks
		Temperature °F	Relative humidity per cent	Temperature °F	Relative humidity per cent	Temperature °F	Relative humidity per cent	
91	Mar. 11, 1912	35.0	100	64.5	92	50.5	93	....
92	Feb. 28, 1912	26.2	64	56.0	94	43.5	90	....
93	Feb. 27, 1912	20.2	85	62.5	89.5	58.7	98.5	....
94	Mar. 7, 1912	26.2	97	63.0	95	63.0	95	....
95	Mar. 4, 1912	27.2	73	56.0	84.5	....	....	This mine has main entry length of only 600 feet..................
97	Mar. 14, 1912	42.0	84	63.2	88.5	53.0	91	....

The necessity for humidifying mine-air to prevent coal-dust explosions is obviated by the high natural humidity of the mines due to the uniform seepage throughout the workings and by the high ash content of the rib-dusts. In only one of the mines examined was steam

FIG. 10. Timbering in haulage entry

exhausted into the air-shaft and in this case it was done to prevent the formation of ice in the shaft in winter. In two of the mines examined, sections of the mine in which the floor had become dry were sprinkled with water by cars occasionally for the purpose of laying the dust.

Table 11 gives the method and frequency of humidification.

TABLE 11—*Method and Frequency of Humidification*

Mine No.	Method of humidifying	Frequency of humidifying
91	Exhaust steam in air shaft	Continuously in winter
92	No humidification	
93	Sprinkling by cars	Twice each week
94	No humidification	
95	No humidification	
97	Sprinkling by cars	Once each month

In the larger mines the use of concrete monolith stoppings is common and the usual proportions of concrete are: 1 Portland cement; 5 unsifted cinders. Large cinders and clinkers are picked out by hand. This concrete is suitable for mine use but it is probable that stoppings could be built more cheaply if concrete blocks were made on the surface and distributed as needed throughout the mine. Table 12 gives costs of various types of efficient stoppings as constructed in Illinois. The efficiency of stoppings depends equally upon the material of which they are constructed and the tightness of the joint between the stopping and ribs, floor and roof. In some mines in the district where sufficiently deep cuts in ribs and floor were not made, leaks have developed due to rashing off of the rib coal and disintegration of the fire-clay floor.

TABLE 12—*Cost and Material of Efficient Stoppings*

District	Material of stopping	Proportions of concrete	Thickness of stopping in inches	Cost in cents per square foot in place: Material	Labor	Total	Size of blocks in inches	Remarks
V or Southern	Concrete blocks	1 cement; 2 sharp washed sand; 4 crushed lime stone	12	10.6	11.0	21.6	8 x 12 x 24	Unnecessary aggregate. Blocks weigh 180 pounds. Labor costs 8 cents per block.
VI	Concrete monolith	1 cement; 6 sifted cinders	6	4.5	6.9	11.4		To replace expanded metal and wood-fibre.
VI	Concrete monolith	1 cement; 2 sand; 5 slack from floor	14			15.0		
VI	Brick coated with cement		4	9.4	7.2	16.6		Brick cost $9.00 per M. delivered. One man builds one stopping, 7 feet by 12 feet, in 2 days
VII	Concrete blocks	1 cement; 6 crushed cinders	8	4.4	6.2	10.6	8 x 8 x 16	Blocks cost 6 cents each at pitmouth
VIII or Danville	Concrete monolith	1 cement; 5 unsifted cinders	8	14.0	11.1	25.1		

The overcasts in the larger mines are often built of brick or concrete and in some instances are lined with ½-inch tongue-and-groove as shown in Fig. 7. Lining the overcast prevents the filling of the air-course in the overcast with an accumulation of small roof-falls which choke the ventilating current.

BLASTING

In only four of the forty-nine mines of the district is coal undercut by machines, but the tonnage of undercut coal totaled 683,789 tons during the year ending June 30, 1912, which amount is 20.2 per cent of the output of the district. Electric chain machines are used exclusively, and the average cut is 6 feet deep. The customary method of placing shots in an undercut face is shown in Fig. 8.

FIG. 11. Destruction of timber by crushing

The tonnage produced by shooting off the solid was 79.8 per cent of the total tonnage of the district for the fiscal year 1912 and, as in other districts, this method of shooting is responsible for great economic waste through increasing the amount of slack coal. The use of powder is excessive and overcharged shots weaken the roof and contribute largely to the accidents from roof-fall. During the year ending June 30, 1912, the average yield in the district was 28.6 tons of coal per 25-pound keg of powder. In the mines shooting off the solid the shooting is done off the weak rib; a round usually comprising four holes: two rib shots and two center shots. The depth of holes as measured in several mines was 6 to 7 feet and the most common diameter was 2¼ inches. The custom of the miners is to put 2½ feet of dummies in a 7-foot hole, leaving in a hole 4½ feet of powder in a bed averaging 5½ feet in thickness.

Black powder was used exclusively at each of the mines examined and was contained in steel kegs; the mines being considered too wet to

allow the use of paper kegs. In this district and also in every district in the State where the steel powder keg is used almost every empty

Fig. 12. Roof-fall in room

powder keg examined had in the head a square hole made by a pick. Familiarity with the use of powder breeds careless handling and it is difficult to prevent the opening of powder kegs with a pick. Even if

Fig. 13. Roof-fall in entry

the pick-hole in the head is not made until after part of the powder has been removed from the keg the danger of igniting the remaining powder

is great. The provision of the State mining law making it a criminal offense for any person to have in his possession in any mine a powder keg with a pick-hole in it should be enforced.

There exists a common disregard of the State law prohibiting the use of "bug-dust" for tamping material and unless shot-firers refuse to fire shots tamped with coal-cuttings the miner ordinarily uses them for tamping because it requires extra labor to obtain fire-clay.

In some mines in the district the powder kegs are placed in a specially constructed covered car for transportation to the working places and are taken into the mine after the men have left the mine and when no current is on the trolley wires. In many of the mines, however, the kegs are carried into the mine on ordinary open pit cars. Recent explosions of powder in the State during delivery to the miners have demonstrated that more care should be exercised to prevent accidents during transportation to the working places.

No fire-runners were employed at any of the mines visited because fires after shots are of very rare occurrence in the district.

Table 13 gives data on blasting practice for each mine examined. The figures for tons of coal gained per keg of powder and per cent of lump coal were obtained from the books of the various companies and are averages for 1910 and 1911.

TABLE 13—*Blasting Practice*

No. of mine	Undercut or shot off the solid	Diameter of holes in inches	Length of holes in feet	Powder fired by	Size of powder	Pounds of powder per ton of coal	Tons of coal per keg of powder	Cost of powder per ton of coal	Per cent of lump over $1\frac{1}{4}$ inches	No. shot firers	No. holes per shotfirer	Remarks
91	Shot off the solid...	$2\frac{1}{4}$	7	Fuse.....	C	1.00	25	$0.07	40	2	125	Average daily output for each miner is seven tons..................
92	Shot off the solid...	$2\frac{1}{4}$	7	..do.....	C	1.00	25	.07	70	2	35	....
93	Undercut..........	$2\frac{1}{2}$	6	..do.....	F	0.21	120	.015	65	4	200	Machine makes six foot cut.................
94	Shot off the solid...	$2\frac{1}{4}$	6	..do.....	C	0.96	26	.067	60	2	150	....
95	Shot off the solid...	$2\frac{1}{2}$	6	Squib....	C	0.57	44	.04	55	0	0	....
97	Shot off the solid...	$1\frac{1}{2}$	8	Fuse.....	C	1.25	20	.088	50	2	60	....

TIMBERING

The treacherous roof of the Danville district necessitates careful timbering in entries and close propping in rooms with the result that the timbering charge per ton of coal mined is unusually heavy. The roof along the permanent haulage-ways falls badly unless supported, and it is often necessary to lag the sets for several hundred feet along the entry when going through areas of bad roof. In the shipping mines timbering of the permanent entries is well done and a great effort is made by the operators to render the haulage entries safe. At each mine examined the three-piece gangway-set was used, either with two long legs, with one leg short and the other long, or with two short legs resting in latches cut in the ribs. Where a curve occurs in the entry the short-

legged frames are commonly used because with long-legged frames there exists the danger of breaking the legs and bringing down bad falls if a trip jumps the track. Fig. 9 shows the methods of leg arrangement in the three-piece gangway set.

Fig. 10 shows typical timbering of a haulage entry, the legs and cross-bars being of 6-inch oak.

The high relative humidity of air in the return and the constant temperature of about 65 degrees are very favorable to sporegrowth and consequently mine timbers decay rapidly and fail. The average life of timber at the mines examined was eighteen months. The gangway-set when weakened by decay usually fails in the cross-bar which is more

FIG. 14. Method of propping near face of room

difficult to replace than are the legs. With a view of avoiding roof falls from breaking cross-bars and of lessening the timbering expense which is constantly increasing on account of the increasing cost and the poor quality of available timber, several mines are substituting steel I-beams for timber cross-bars, but retaining timber legs in the permanent entries. The standard I-beam of structural steel which combines a high degree of resistance to bending with minimum weight of metal has proved well fitted for use in mines. In the Danville district there are the following relations between sizes of steel I-beams and length of span:

Span in feet	Size of I-beam in inches
8	4
10	6
12	8
16	12

With this system frames are spaced on 2½-foot centers. Eight-inch diameter rough white oak legs are used with spans of 8 and 12 feet, and 10-inch legs are used for greater spans.

Old railroad rails have been used as cross-bars in a few mines. In one mine using 60-pound rails as cross-bars the rails failed under the roof weight on account of the crystalization of the rail. Rails break easily under bending stress because of the high carbon content of the steel from which they are made.

It is an open question whether or not the use of white oak legs with steel I-beam cross-bars is an economy in permanent timbering. The

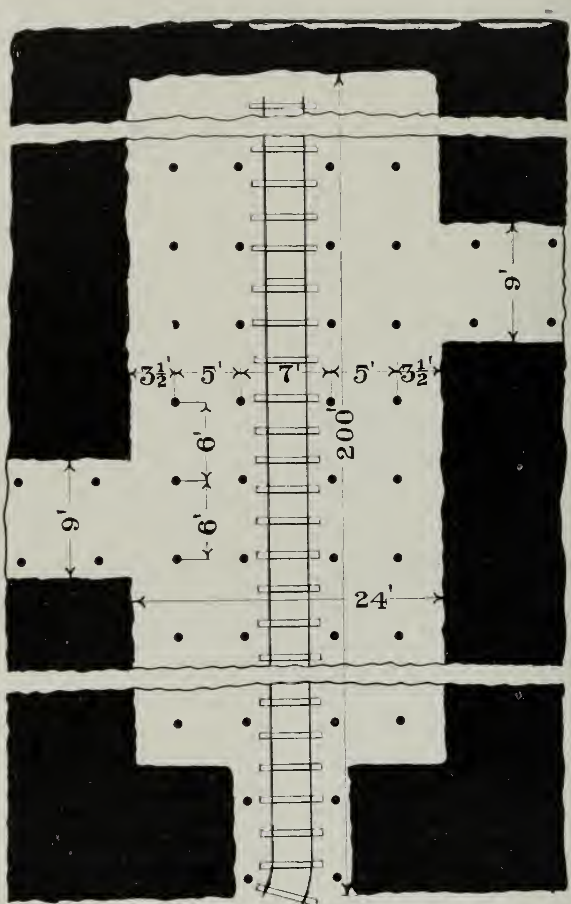

Fig. 15. Plan of room, showing propping

first cost of a 4-inch 28-pound steel leg 6 feet in length would be approximately $4.00, delivered in the mine. An 8-inch round white oak leg 6 feet long costs, delivered in the mine, approximately 50 cents. To the cost of replacement of white oak legs during the life of the mine must be added the cost of fire hazard and increased insurance rate.

None of the mines examined attempted to give preservative treatment to mine timber, because under the unusual roof conditions

obtaining in the district it is not regarded as economy to apply preservative treatment which equals at least the first cost of the timber to

FIG. 16. Steel I-beams and brick pillars at bottom

material liable to destruction by crushing, as shown in Fig. 11, within a few months after its installation.

FIG. 17. Inby end of concrete-lined bottom

The greatest menace to the life of the miner in this district is the treacherous roof, and consequently adequate propping in rooms is compelled by the larger operating companies; but in the mines of small

capacity discipline is more lax, and the miner is not compelled to keep his props close to the face. There is danger in failure to keep the props within 12 or 15 feet from the working face in a district where it is often necessary to have cross-bars across the track and along the ribs, yet it is only by constant diligence on the part of officials that the miners can be made to keep their places safe. The necessity for close propping is greater in this district than in many others in the State because the numerous "nigger-heads" or "sulphur-balls" which protrude from the roof have little cohesion to the roof shale. The accident record of the district shows the need of greater vigilance on the part of mine officials and greater regard for their own safety on the part of the miners. The district during the year ending June 30, 1912, produced 5.8 per cent of the coal mined in Illinois and the number of days work performed was 5.8 per cent of the days work performed in coal mines in the State.

FIG. 18. Concrete and steel-I-beam in slope bottom

It should therefore have furnished 5.8 per cent of the accidents occurring in the State, but with fifty-nine non-fatal accidents it supplied 7.4 per cent of the non-fatal accidents and with eighteen fatal accidents it supplied 10 per cent of the fatal accidents. Of the non-fatal accidents in the district 47.5 per cent were caused by falls of rock and coal as compared with 45.5 per cent from this cause for the State as a whole. Of the fatal accidents 77.7 per cent were caused by falls of rock and coal as against 54.4 for the State. Figs. 12 and 13 show typical roof falls in the district.

At each mine examined the props were counted in place in a measured length in each of several typical rooms. The width of these rooms was measured and there was thus obtained the average number of props per hundred square feet of roof. For each mine visited the number of props per 100 square feet of roof and other data concerning propping are given in Table 14. The figures referring to the number

and cost of props per ton of coal were obtained in each case from the books of the operating company. Fig. 14 shows the face of a typical room in one of the larger mines. Here, props are placed within 12 feet of the face, allowing only sufficient space between the last prop and the face for the operation of an undercutting machine. Fig. 15 shows a plan of a typical room in a mine where adequate propping is insisted upon.

The roof of the "shaft bottom" in nearly all the larger mines in the district is supported by concrete or steel. Fig. 16 shows the arrangement of a bottom where brick pillars 13 inches by 18 inches set on 8-foot centers support 4-inch steel I-beams on which are laid 2-inch by 6-inch planks for lagging. The pillars are 7 feet high. This arrangement gives a bottom free from the danger of roof-falls but not fire-proof.

Fig. 17 shows the inby end of a concrete bottom. The walls of the lining of the entry are 24 inches thick at the bottom and the thickness of concrete is gradually reduced till at the top of the arch it is 12 inches. A gob filling is packed between the arch and the roof. The length of concrete bottom on each side of the shaft is 165 feet. The concrete was made in the following proportions: 1 Portland cement: 1 sand: 4 washed gravel.

TABLE 14—*Data Concerning Props in Rooms*

No. of mine	Number per 100 square feet of roof	Cost in cents per 100 square feet of roof	Diameter in inches	Length in feet	Life in months	Distance in feet of nearest prop from face	Per ton of coal	
							Number	Cost in cents
91	3.1	20.9	5	6	18	16	0.25	1.7
92	8.1	44.6	6	5½	18	15	0.31	1.7
93	3.1	24.8	5	8	18	18	0.18	1.2
94	7.6	41.8	4	5½	18	11	....	....
95	5.0	37.5	6	7½	18	25	....	....
97	6.3	44.1	4	7	24	14	0.33	2.3

Fig. 18 shows a slope bottom where concrete is placed on the top of coal which is 5 feet 9 inches thick at this point. The wall is 2 feet high and 9 inches thick and the concrete has the proportions: 1 Portland cement; 6 river-washed gravel. Ten-inch steel I-beams, weighing 40 pounds to the lineal foot, are placed on 2½-foot centers across the caging-room to support the roof, and the space between the I-beams is filled with concrete, thus providing a water-proofed roof. It is doubtful if placing concrete on top of the coal will be successful, for if the coal spalls off rapidly this arrangement will not be permanent.

In the lining of shafts and slopes in the district timber is principally used. Nearly all the shafts were sunk before the State law requiring fire-proofed shafts was passed.

Fig. 3 shows a concrete-lined slope which is 128 feet long from the surface to the bottom, 7 feet wide, and 8 feet high. The concrete sides are 6 feet high and 9 inches thick. Placed on 2½-foot centers along the concrete wall are white oak legs 6 inches in diameter and 18 inches long which support 56-pound rails on which the lagging rests.

HAULAGE

Few mines in the district have a daily production of over 1,500 tons, and mules are generally used on both main and secondary haulage. Six mines are provided with electric locomotives on the main haulage and in one mine haulage is done by cable on the main entries. The bad condition of the secondary haulage roads makes the use of gathering locomotives unprofitable and although they have been tried in this district their use has been discontinued and mules are universally used for gathering. A decrease in the cost of mining in this district can probably be made by bettering the condition of the haulage-ways. More attention advantageously could be paid to maintaining easy grades and curves. As beds 6 and 7 lie practically flat, easy grades can be maintained with

FIG. 19. Typical caging-place at shaft-bottom

very little brushing of roof and floor. The tendency of the floor to heave is mainly caused by excess weight on pillars of insufficient size. The present operators of the mines of large output did not project the mines and are, therefore, not responsible for the poor condition of the haulage-ways, and a successful effort is being made in some mines to improve the uneconomical haulage.

Rails in the main haulage often weigh only 16 pounds and 16-pound rails are used on the secondary haulage at all mines. The track-gage varies from 36 to 38 inches. At each mine examined wooden rails are used in the rooms. In the shipping mines ties are usually 5 inches by 6 inches by 5 feet and are generally of white oak. Pit cars often are not kept in good repair, and the constant dropping of coal from these

FIG. 20. Abandoned drag-line excavator

cars to the track and the presence of gob along the track in many instances hinder rapid and cheap haulage.

The shaft bottoms in the district are too short for the most economical handling of the output but the short bottom is a product of an earlier period and is found throughout the State. Table 15 gives haulage data for the six shipping mines examined.

FIG. 21. Island of overburden of stripping mine

With the widening of haulage entries in dangerous places since the present operating companies took over the large mines, the percentage of fatal accidents due to pit cars has decreased till in 1912 the per cent of fatal accidents from this cause was 5.5 for the district, as against 18.8 for the State, and the per cent of non-fatal accidents 27.1 for the district, as compared to 26.3 for the State.

TABLE 15—*Haulage Items*

No. of mine	Kind of haulage on main-entries	Weight of mine cars in pounds	Capacity of cars in tons	Main-entry rail weight in pounds per yard	Track gage
91	Mule	1,800	$2\frac{1}{2}$	30	38
92	Mule	1,150	$1\frac{1}{2}$	16	38
93	Electric locomotive	2,200	$2\frac{1}{2}$	30	38
94	Main and tail-rope	1,800	$1\frac{1}{4}$	16	36
95	Mule	1,200	1	16	38
97	Mule	1,200	1	16	36

HOISTING

Inasmuch as all of the mines of this district are of moderate daily output and therefore do not require uninterrupted hoisting at high

FIG. 22. Removing top-soil with hydraulic monitors

speed, the hoisting equipments are not remarkable in any respect. The hoisting shafts are all timber lined and of moderate size, ranging from a single compartment, 5 feet by 8 feet, to double compartment, 9 feet by 14 feet. The hoisting engines are usually 16 inches by 32 inches, only two of the mines examined having 24-inch by 36-inch engines. Cable drums are of ordinary size, varying in diameter from 5 to 7 feet.

Fig. 23. Shale and top-soil overlying coal

Many of the mines were opened before the perfection of automatic caging and at none of the mines examined was automatic caging practiced. This district played an important part in the development of the self-dumping cage and all of the larger mines use this type of cage. Table 16 gives data concerning hoisting equipment for each mine examined.

Fig. 19 shows a caging place typical of this district.

Table 16—*Equipment for Hoisting*

No. of mine	Coal reached by—	Average daily tonnage	Hoisting shaft		No. hoisting compartments	Self dumping cage	Hoisting engine		Drum	
			Depth in feet	Size in feet			Type	Size in inches	Diameter in feet	Length in feet
91	Shaft....	1,250	217	8 x 12	2	Yes....	Direct connected.	24 x 36	7	2½
92	Shaft....	150	240	7 x 12	2	No.....	..do............	16 x 32	5¼	4
93	Shaft....	2,600	186	9 x 14	2	Yes....	..do............	24 x 36	7	2½
94	Shaft....	800	90	6 x 14	2	..do...	..do............	16 x 32	5	2
95	Slope ...	50	Slope	5 x 8	1	No.....	Horse whim	....	5	4
97	Shaft....	300	223	......	2	Yes....	Direct connected.	16 x 32	5	4

PREPARATION OF COAL

There are no coal washeries or rescreening plants in the district and all sizing is done over gravity-bar or shaking screens. The separation into sizes goes no further at the average mine than to divide the output into coal called lump that will go over bars spaced $1\frac{1}{4}$ inches apart and coal called screenings that will go through these bars. The mines of largest output at times make a further separation dividing the coal which goes through the $1\frac{1}{4}$-inch screen into the three following sizes:

Name	Size
Screenings	Over 1 inch and under $1\frac{1}{4}$ inches
Nut	Over $\frac{3}{8}$ inch and under 1 inch
Slack	Under $\frac{3}{8}$ inch

Fig. 24. Steam-shovel digging shale overburden

A very small per cent of the total tonnage of the mines is shipped as run-of-mine; the largest amount at any mine being 25 per cent of the output. A few of the mines makes a small amount of 3-inch and 4-inch lump. A cleaner separation into the required sizes can be obtained at several mines by the installation of a longer screen. Table 17 gives data concerning tipple equipment. The tipples of the district are all of frame construction.

As stated on page 12 iron pyrites or "sulphur" occurs more plentifully in bed 7 than in bed 6, and when the pyrites is not disseminated through the coal but is present in nodular form, it can easily be separated from the coal by hand at the working face. This separation serves the double purpose of making cleaner coal and of segregating a valuable by-product. The pyrites thus obtained was sufficient in quantity at two mines working in No. 7 bed to warrant the erection of small plants for removing the coal that adheres to the nodules of pyrites before the pyrites is shipped to a sulphuric-acid plant. The pyrites with adhering coal is crushed to $1\frac{1}{2}$-inch mesh and elevated to a bin whence it is dis-

FIG. 25. Drilling hole for blasting at stripping-mine

charged into a revolving trommel 4 feet long and 3 feet in diameter with 2-inch round holes. The oversize from this trommel goes to a one-cell jig for washing. The undersize goes to a second trommel with $\frac{1}{4}$-inch perforations and the undersize from the second trommel is discharged into a three-cell jig which separates coal and pyrites. The oversize from the second trommel and the undersize from the first trommel are elevated to a third trommel with $\frac{5}{8}$-inch perforations from which the oversize goes to market and the undersize to the three-cell jig which cleans the fine pyrites.

FIG. 26. Dinky locomotive and haulage-way at stripping mine

FIG. 27. Stripping mine haulage way

The surface power plants of this district present no unusual features either of arrangement or size. Fig. 2 shows a typical surface plant. The total horsepower developed at any mine is not large and on

FIG. 28. Stripping mine through-cut

account of the moderate outputs there is no necessity for the storage of many empty cars above the tipple. Table 18 gives data on surface equipment for each mine examined.

TABLE 17—*Tipple Equipment*

No. of mine	Coal bed	Screen				Per cent of coal over $1\frac{1}{4}$ inches
		Type	Length in feet	Width in feet	Inclination in inches per foot	
91	6	Bar	12	8	4	40
92	6	Bar	24	8	$4\frac{3}{4}$	70
93	6	Bar	33	9	4	65
94	7	Shaker. 100 shakes per minute	22	6	4	60
95	7	Bar	6	4	4	55
97	7	Shaker. 100 shakes per minute	36	8	4	50

TABLE 18—*Surface Equipment*

No. of mine	No. loading tracks under tipple	No. empty cars possible to store above tipple	Boilers				Electric generators		
			No.	Type	Total horse power	Average steam pressure	Manufacturer	K. W.	Voltage
91	2	60	2	Fire-tube	300	120	Westinghouse	10	250
92	2	8	1	Fire-tube	90	125			
93	4	57	6	Fire-tube	900	100	Westinghouse	150	275
94	2	24	3	Fire-tube	450	100	Western Electric	$22\frac{1}{2}$	250
95			1	Water-tube	18	80			
97	4	50	3	Fire-tube	310				

FIG. 29 Stripping-mine coal face

STRIPPING

In the Danville district the removal of overburden from coal lying at depths of 20 to 30 feet below the surface has been practiced for a longer time and more extensively than it has in any other district in Illinois. Beginning in 1866 with the primitive method of exposing the coal by removing a very shallow overburden by means of scrapers dragged by horses the process developed slowly, horse scrapers being replaced about 1900 by the drag-line steam-shovel, shown as it appears at the present time in Fig. 20, which, in turn, was replaced by the revolving steam-shovel in 1910.

The methods of stripping now employed in the district differ in the path which the shovel follows while digging, in the manner in which the top soil is removed from the shale overlying the coal, and in the disposal of the spoil. In one method the shovel makes a continuous cut about 50 feet wide in a circle around the area to be stripped and the coal

Fig. 30. Stripping mine spoil-bank

exposed behind the shovel is mined. The shovel having completed the first circle begins a second just inside the first and continues to move in circles with constantly decreasing diameters. Fig. 21 shows the island of overburden around which the shovel travels when stripping by this method.

In the second method, the shovel instead of traveling in a circle, goes forward and back across the property in parallel straight lines, a haulage-way for disposing of the material mined being maintained at one side of the property.

Where the shale overlying the coal is to be used for the manufacture of brick or for other purposes and is overlaid by top soil, the soil is first removed by hydraulicing. As fast as the coal is mined, the top soil for

Fig. 31. Steam-shovel with belt-conveyor spoil-elevator

50 feet from the edge of the bank of the cut is washed off into the pit left by the removal of the coal. For washing off the top soil hydraulic monitors under a pumping-head are used as shown in Fig. 22. The amount of top soil washed per eight-hour shift varies with the material

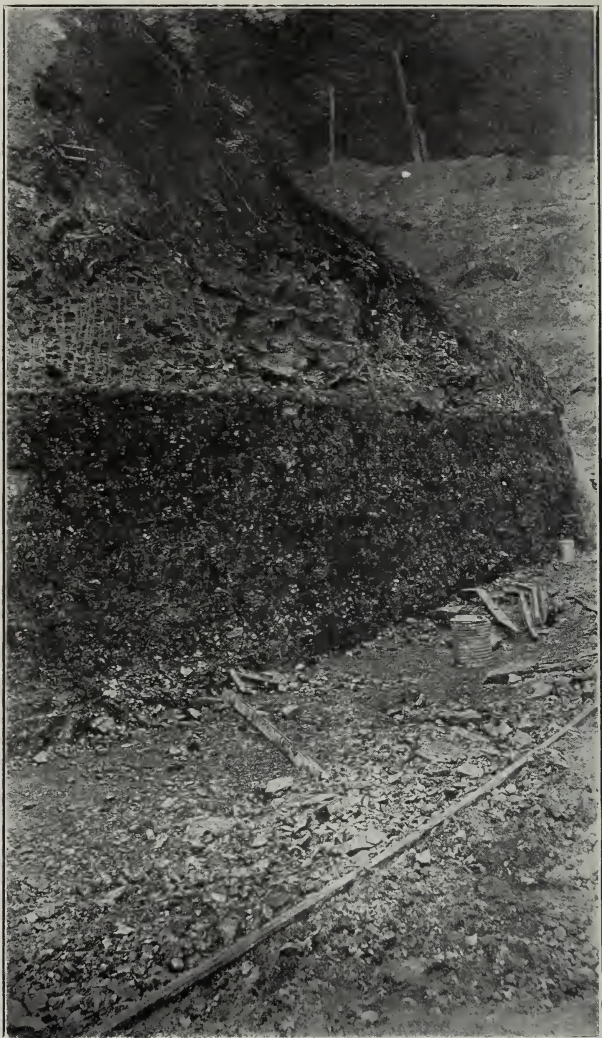

FIG. 32. Section of bed No. 7

removed; in tight ground 100 cubic yards may be the total for the eight hours; in loose ground 2,000 cubic yards may be washed off. Fig. 23 shows in the foreground the soil already washed down into the pit and in the background the overburden of shale and soil overlying the coal.

Fig. 24 shows the shovel digging the shale overburden and exposing the coal after the top soil has been washed off. Fig. 25 shows the method of drilling holes for blasting the coal. In blasting stripped coal, holes 2½ inches in diameter are drilled 12 feet apart at a distance of 5 to 9 feet from the face. The average charge of powder is 1½ pounds per hole, and the average gain per 25-pound keg of powder is 100 tons. At this stripping mine the hauling is done by small steam locomotives as shown in Fig. 26.

At one stripping mine in the district the steam-shovel digs a permanent haulage-way along one side of the area to be stripped as shown in Fig. 27. At the end of this haulage cut a thorough-cut about 50 feet wide is made along the boundary of the property. Fig. 27 shows the branching of the thorough-cut from the haulage-way and Fig. 28 shows the shovel at work in the thorough-cut. The exposed coal is mined behind the shovel as is shown in Fig. 29. When the thorough-cut reaches the property line the shovel turns around and digs the overburden from another strip about 50 feet wide depositing the spoil in the pit made by the removal of the coal exposed by the "thorough-cut." This spoil bank is shown in Fig. 30.

Fig. 31 shows a steam-shovel which elevates the spoil by a belt-conveyor and deposits it along the side of the shovel-cut.

The tipples at stripping mines are usually rough housings of shaking-screens raised sufficiently above the railroad tracks to allow dumping into cars and approached by a steep incline up which the pit cars are drawn by cable.

Fig. 32 shows a section of coal No. 7 in this district and also shows the shale and top soil overlying the coal. The total cost of mining coal by stripping the overburden varies on a daily output of 300 tons per day from 40 to 50 cents per ton loaded on the cars.

PUBLICATIONS OF THE ILLINOIS COAL MINING INVESTIGATIONS

Bulletin 1. Preliminary Report on Organization and Method of Investigations, 1913.

Bulletin 2. Coal Mining Practice in District VIII (Danville), by S. O. Andros, 1914.